Wisconsin Geological
and Natural History Survey

I. C. 6327

SEPTEMBER, 1930

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT THE CONSOLIDATED CORTEZ SILVER MINE, CORTEZ, NEVADA

BY

GEORGE. W. HEZZELWOOD

Digitized by Google

I. C. 6327
September, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT THE CONSOLIDATED CORTEZ SILVER MINE, CORTEZ, NEVADA[1]

By George W. Hezzelwood[2]

INTRODUCTION

This paper describing the mining methods at the mine of the Consolidated Cortez Silver Mines, Cortez, Nevada, is one of a series of similar papers being prepared by the Bureau of Mines on mining practices and costs in the various mining districts of the United States.

The Cortez district is situated on the southwest slope of Mt. Tenabo of the Cortez Range, near the middle of the boundary line between Eureka and Lander Counties (fig. 1). The camp of Cortez is 36 miles south of Beowawe, the nearest station on the Southern Pacific and Western Pacific Railroads. Mt. Tenabo, which is the highest summit of the south end of the Cortez Range, has an elevation of 9,240 feet; Cortez is about 6,280 feet above sea level. The Consolidated Cortez Silver Mines is the only company operating in the district at the present time and employs about 60 men. The daily production of 125 tons of silver-lead ore is concentrated in the company's mill of 150 tons capacity. The concentrate is hauled to Beowawe by automobile trucks for shipment to a reduction plant in Salt Lake Valley, Utah.

ACKNOWLEDGMENT

The author acknowledges the assistance of William O. Vanderburg, associate mining engineer, U. S. Bureau of Mines in the preparation of this paper.

HISTORY

The Cortez district was discovered in 1863; rich ore yielding several hundred dollars per ton was hauled by pack train to Austin, 60 miles away, for treatment. In 1864, an 8-stamp mill was erected in Mill Canyon, 4 miles from Cortez, on the north slope of Mt. Tenabo. In 1867, Wenben, one of the original

---

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6327."

2 General superintendent, Consolidated Cortez Silver Mines, and consulting engineer, U. S. Bureau of Mines.

Digitized by Google

locators, acquired by purchase all the important claims on the Cortez side of Mt. Tenabo and organized the Mt. Tenabo Mill and Mines Co. In 1869, Wenben purchased the Mill Canyon mill which had been enlarged to 16 stamps. A new mill was constructed at Cortez by the Tenabo Mill and Mines Co. in 1886, the treatment process being a chloride roast followed by leaching with sodium hyposulphite. From 1889 to 1891, the Tenabo Mill and Mines Co. was operated by the Bewick-Moering Syndicate under the name of the Cortez Mines (Ltd.); the property was turned back to Wenben in 1892. Wenben died in 1895 and only a small amount of work was done by lessees from the time of his death until 1920, when the property was purchased by the Consolidated Cortez Silver Mines Co. This company completed the erection of a 125-ton cyanide plant in 1923, which operated continuously until the later part of 1927, when the mill was remodeled and converted into a 150-ton flotation plant.

PRODUCTION

The production of the Cortez and Mill Canyon districts from the date of discovery to the year 1910, is estimated by Emmons[3] to be $10,000,000, most of which was taken from the Cortez mine. The Consolidated Cortez Silver Mines Co. from 1919 to the end of 1929 has produced 267,249 tons of ore, the net value of which was $2,036,892. Of this amount 138,905 tons was taken from old fills on the upper levels of the mine. This ore had been too low grade to work profitably until the modern milling plant was installed. The balance of 128,344 tons was mined by the shrinkage system from the lower levels of the mine. Silver is the principal product derived from the ore.

GEOLOGY

The geology of the Cortez district has been briefly described by William H. Emmons.[4]

The formation consists of a series of limestone beds and quartzites which have been uplifted by an intrusion of granodiorite and later cut by a series of dikes (fig. 2).

Three types of ore deposition have been observed:

(1) Dike ore, occurring in and immediately adjoining the dike fissures.

(2) Bedding plane ore, occurring along the bedding planes of the limestone.

(3) Fissure ore, occurring in fissures approximately parallel to the dikes. This type of ore deposition has furnished the major portion of the production.

---

3 Emmons, William H., A Reconnaissance of Some Mining Camps in Elko, Lander and Eureka Counties, Nevada: U. S. Geol. Survey Bull. 408, 1910, 130 pp.

4 See footnote 3.

Digitized by Google

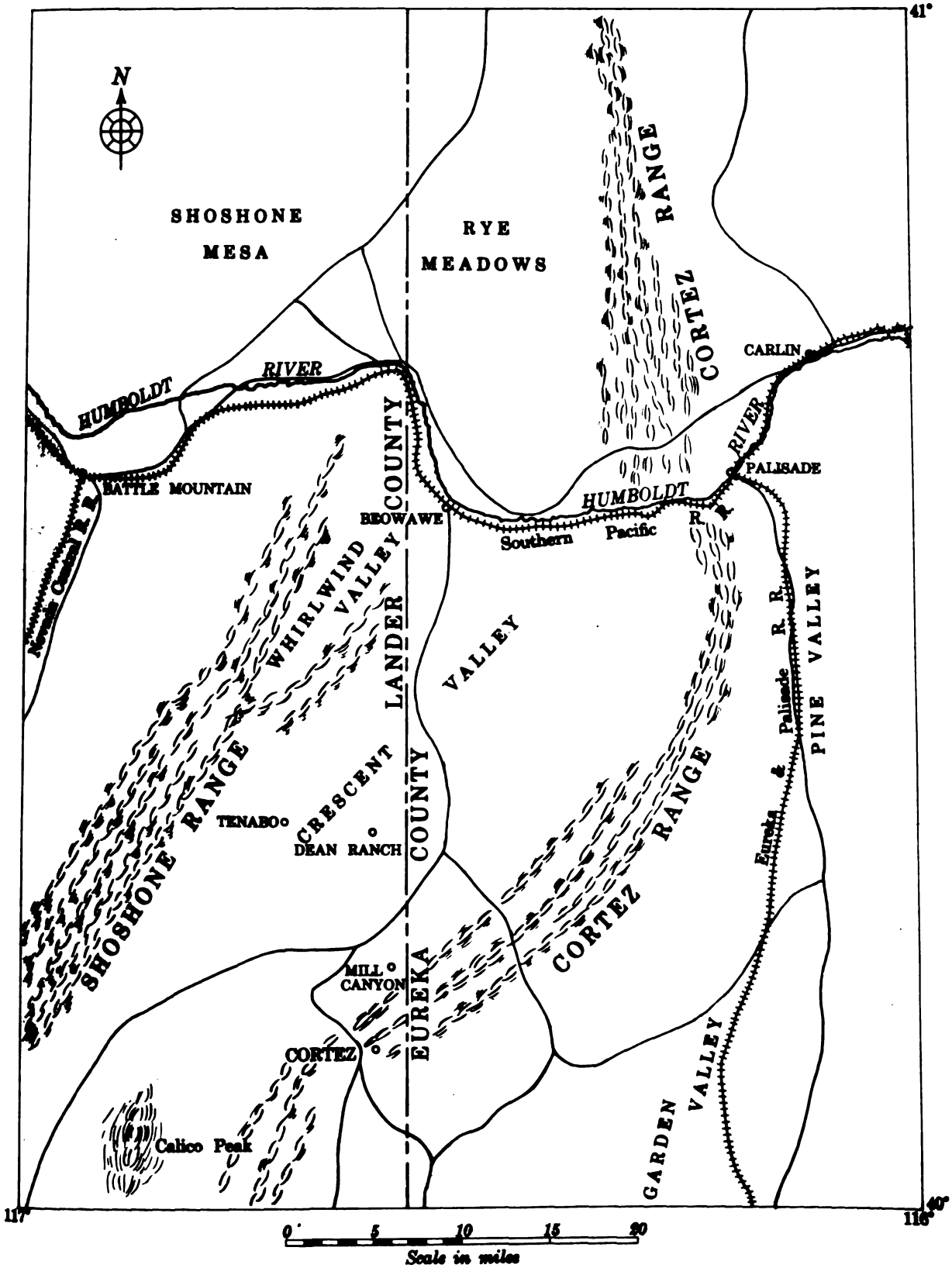


Figure 1. – Map of territory around Cortez, northeast Nevada

Digitized by Google

The fissure ore bodies vary from 1 to 20 feet in width; the average width is about 7 feet. The maximum length of the ore bodies is 600 feet. The dip varies from 45 to 80°, and no difficulty is encountered in drawing ore from the stopes by gravity. The walls are poorly defined, being economic rather than structural.

The walls of the fissures are invariably solid and consequently well adapted to the shrinkage method of mining. Very little support is necessary in the stopes. In places the ore breaks in large boulders; these boulders are broken in the stopes with hand hammers or plugged by means of jackhammers and blasted.

The valuable minerals in the unoxidized ore bodies consist of silver-bearing quartz, galena, pyrite, tetrahedrite, sphalerite, and chalcopyrite.

PROSPECTING AND EXPLORATION

Indians gathering "float" in the gulches at the southern end of the Cortez Mountains in the early sixties led prospectors to investigate the cliffs above the places where the float ore was found. Inspection of the cliffs resulted in the discovery of surface croppings of ore, which was followed downward by an inclined shaft. Subsequent exploration for ore bodies has been done by a combination of two methods:

(1) By a series of adits driven to intersect the projected positions of the ore deposits at different elevations below the outcrop.

(2) By inclined shafts driven on the foot walls of the ore bodies from the adit levels and by lateral drifts driven at intervals from the shafts. When the nature of the topography made exceptionally long adits necessary to reach the ore bodies this method was adopted.

In the portion of the mine which is being worked at present, three more or less distinct fissure ore zones have been outlined. Careful study is made of the minor structural features, and wherever vein quartz is found within these zones it is followed and almost invariably leads to commercial ore bodies. Driving cross-cuts to the projected positions of ore bodies or structural features has not been successful. A general rule that has been adopted for carrying out development work is to follow the quartz. When an ore body is found its limits are determined by work in the ore. The search for ore has been facilitated by experience gained in previous work, and as development progresses much geological information is gained which helps in future work, namely, determining the rake of the ore bodies, noting the relation of the ore-bearing fissures to the main dikes, and observing the effect of cross fracturing on ore disposition.

METHODS OF SAMPLING AND THE ESTIMATION OF ORE RESERVES

All headings progressing in ore are sampled after each round blasted. Samples are taken of all cross fractures and recorded for future reference. Samples are taken with a hand pick across the width of the ore; they are useful more as a record of the presence of ore than as a basis for estimating ore tonnages.

Digitized by Google

While stoping is being done the chutes are sampled during each shift. Shift bosses take as many stope samples as may be needed to outline the ore and as factors for estimating the grade of ore broken in the stopes. The chute samples are taken for the purpose of regulating the mill feed, both as to values and as to sulphide and oxide character.

The usual methods of blocking out the ore by measuring and sampling in making estimates of ore reserves has been found unreliable at the Cortez mine. The tendency of the ore to narrow or widen and the grade to change without apparent reason makes such methods inaccurate. A ratio between the number of feet of development work and the number of tons mined has been worked out for the operations on the lower levels which were started in 1926. This ratio furnishes a basis for estimating probable ore, particularly when development work is confined to the three known zones. This method of estimating, although not accurate, is probably as safe as any method for this form of ore deposit.

DEVELOPMENT

Entry to the part of the mine being worked at present is made by two adits and one inclined shaft from the lower adit. The portal of the Garrison, or upper adit, has an elevation of 7,055 feet; and the Arctic, or lower adit, is 6,701 feet above sea level. The shaft which has an inclination of 73° extends 240 feet vertically below the Arctic level. From this shaft two levels have been established, which are respectively 98 feet and 217 feet vertically below the level. (Fig. 2.)

Shaft

The shaft is 6 by 12 feet in rock section and is untimbered except for bearing stulls for the skip track (fig. 3). The bearing stulls 8 to 10 inches in diameter are wedged into hitches cut in the sides of the shaft on 5-foot centers. Thirty-pound rails are spiked to the bearing stulls.

The shaft was recently sunk 120 feet to open up a new level. Drilling was done on one shift and mucking on the other. A heavy bulkhead of round timbers with a trap door was put in the shaft below the skip pocket on the first shaft level. Extension rails were used so that the skip could be lowered to the bottom of the shaft. Ore was hoisted from the first level during the shift when drilling was being done in the shaft. The main skip was used on the other shift to handle the rock broken in the shaft sinking; the rock was shoveled directly into the skip.

Blasting was done with delay electric detonators. The drill crew consisted of two miners. Drilling was done with jackhammer machines using 7/8-inch hexagon hollow steel. The shoveling crew consisted of four men. Timbering was done while drilling was in progress. A small air hoist was used to lower timber, steel, and drilling equipment to the bottom of the shaft. Figure 4 shows the number of holes, order of firing of holes, and amount of 40 per cent strength gelatin dynamite used in the average shaft round.

Digitized by Google

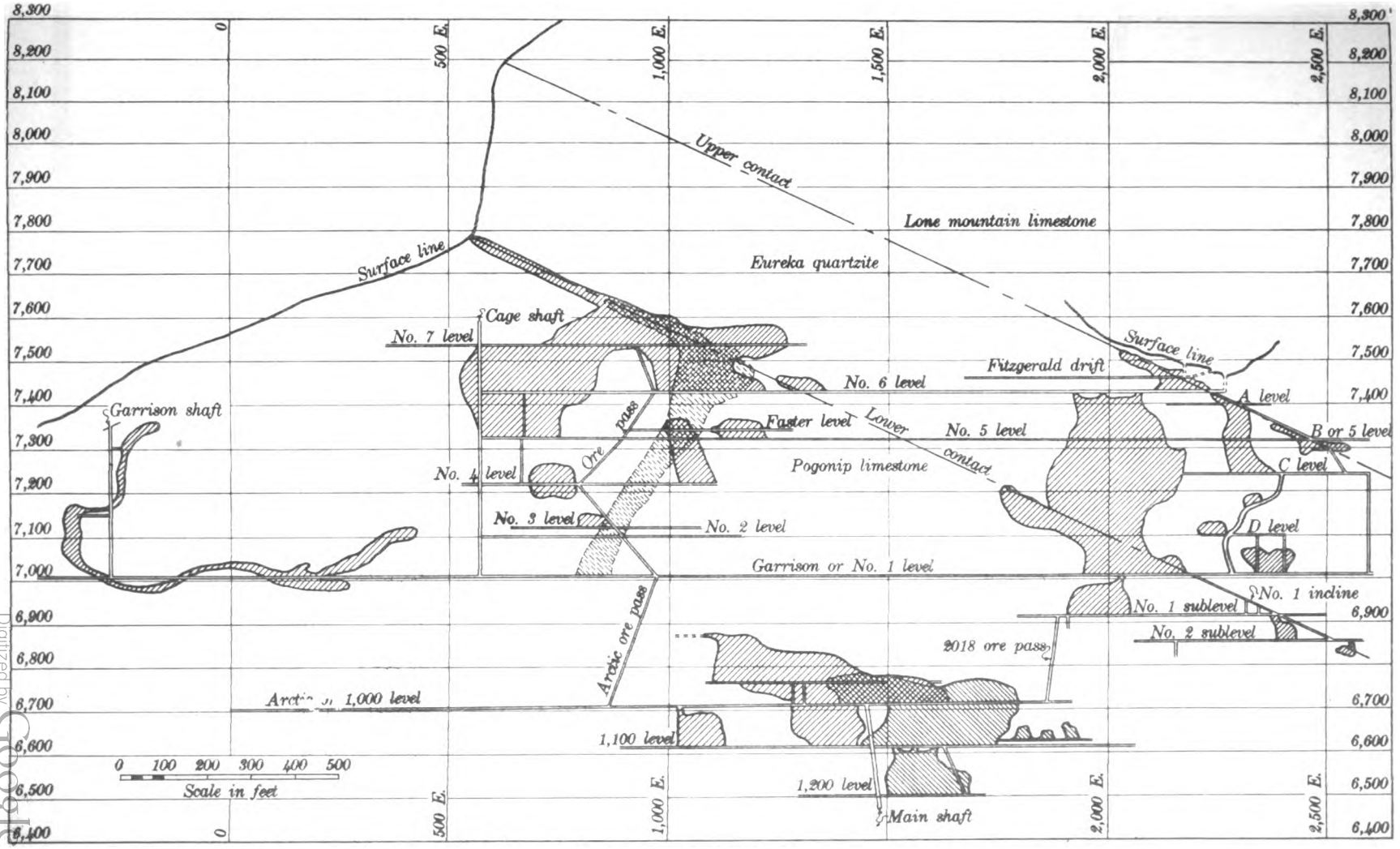


Figure 2. – Longitudinal sections through No. 1 dike showing development methods. Cross hatching indicates stoped ground on or adjacent to No. 1 dike

Digitized by Google

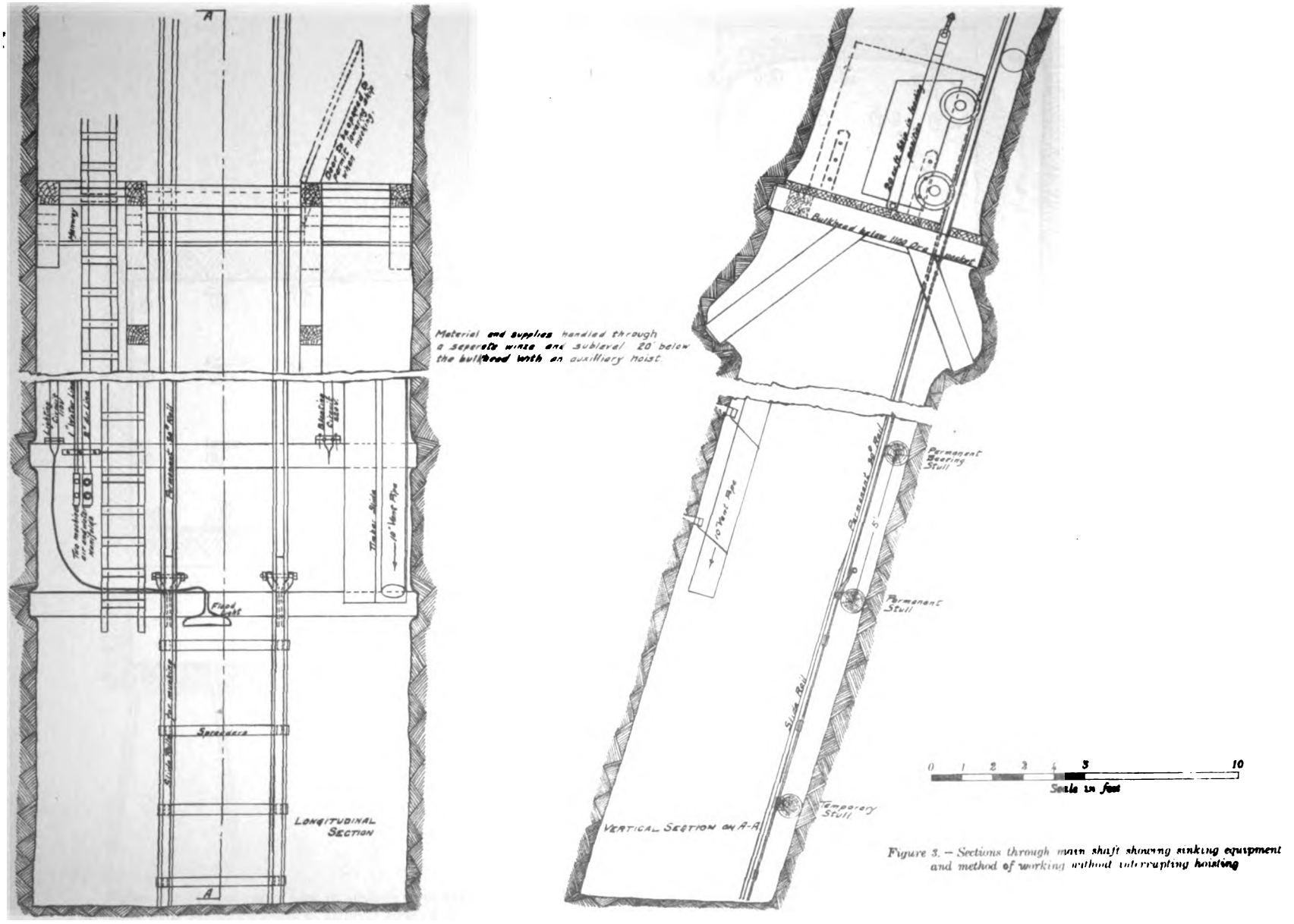


Figure 3. — Sections through main shaft showing sinking equipment and method of working without interrupting hoisting

Digitized by Google

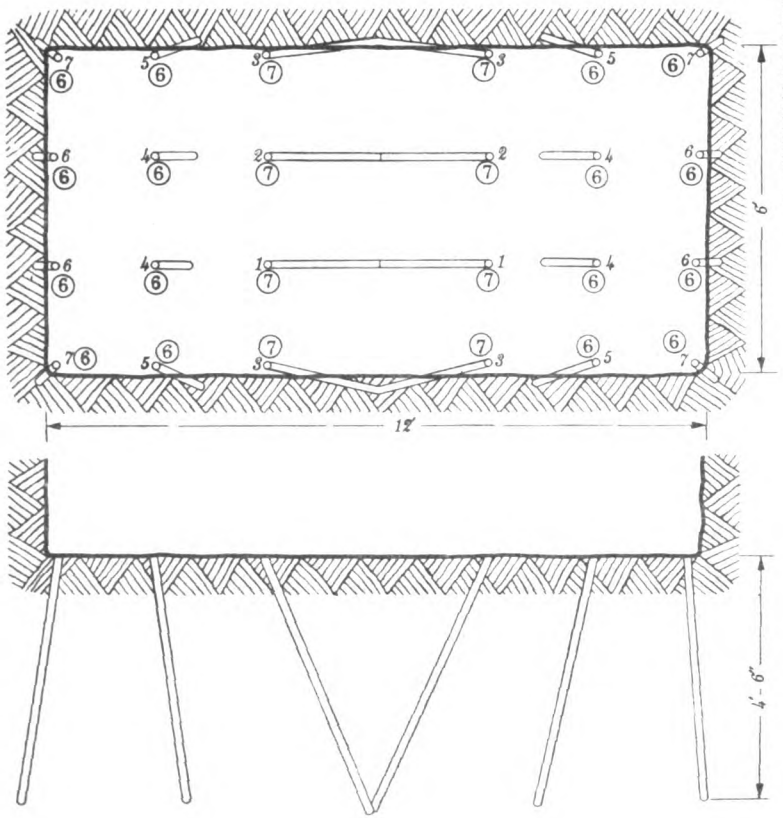


Figure 4. – Type of round used in sinking main shaft. Numbers indicate order of firing. Numbers in circles cartridges of 40% powder used

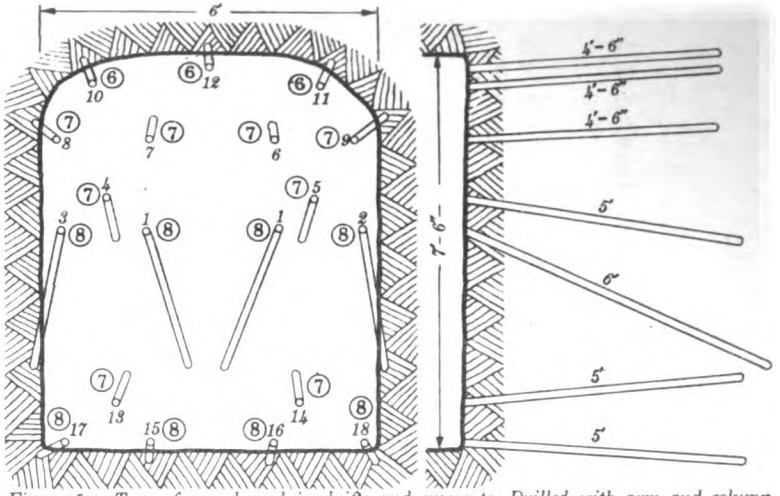


Figure 5. – Type of round used in drifts and crosscuts. Drilled with arm and column. Numbers indicate order of firing. Numbers in circles cartridges of 40% powder used

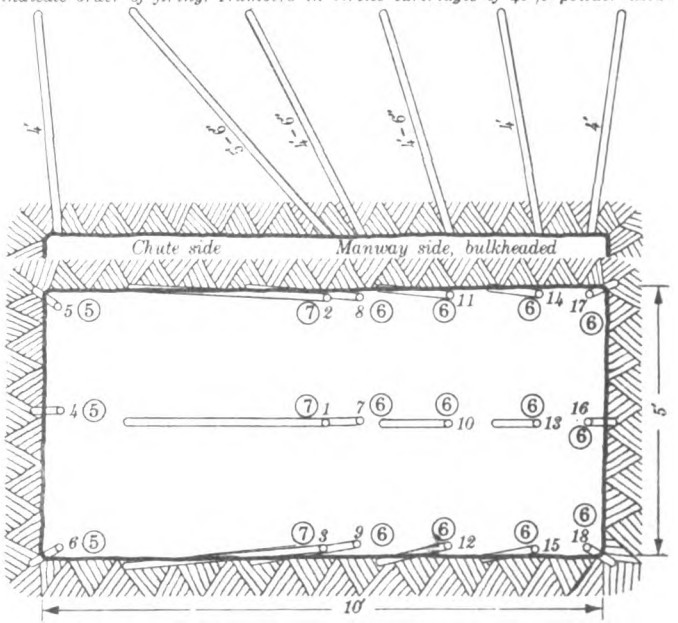


Figure 6. – Type of round used in manway raises. Drilled with stoper machines

Digitized by Google

Adits, Drifts, and Crosscuts

All lateral headings are untimbered and average 6 feet wide by 7½ feet high.

The approximate routine of operations in advancing an adit, drift, or crosscut when drilling is done on one shift is as follows:

Day shift (1 miner)		Night shift (2 muckers)	
Operation	Time	Operation	Time
Barring down	15 minutes	Barring down	30 minutes
Setting up machine	30 minutes	Shoveling and tramming	7 hours
Drilling	5 hours		
Tearing down	20 minutes		
Preparing primers and loading holes	30 minutes		
Blasting	10 minutes		

Headings are driven with Leyner machines mounted either on an arm attached to a column or directly on a crossbar. The crossbar is used when speed is desirable so that drilling and mucking are done at the same face simultaneously. One and one-quarter inch hollow round shanked sheel is used. Starters are 2⅓ feet long with a bit gage of 2½ inches. The gage is reduced one-eighth of an inch for each 2 feet change in length. The last steel is 6½ feet long.

Figure 5 shows the type of round used in drifts and crosscuts. Forty per cent strength gelatine dynamite is used with No. 6 detonators and 8-foot lengths of single-taped fuse. From 85 to 100 sticks of dynamite, size 1-1/8 by 8 inches, are required per round, depending on the character of the ground.

Drill steel and blasting supplies are brought to the working places by the tool nipper. Pipes and track are installed by a crew of two men. All shoveling is done by hand. On the lower levels tramming is done by hand; on the main haulage level a locomotive handles the cars from the development faces.

Raises

Three types of raises are used: An ore-pass raise to transfer ore from the upper to the lower adit level; manway raises to afford access to the stopes for men and supplies; and chute raises, through which the ore is drawn from the shrinkage stopes into the cars.

The ore-pass raise was driven from the lower adit level to the upper workings a distance of 274 feet. This raise was carried 5 by 10 feet in rock section and inclined 80° from the horizontal. Dividing stulls were put in on 5-foot centers, and these stulls were lined on the chute side with doubled 2-inch plank lagging in the lower half of the raise and 3-inch lagging in the upper half. The second compartment of the raise was used as a manway.

Digitized by Google

In advancing this raise the dividing stulls were placed in position close to the back after each round for the first 150 feet. A temporary bulkhead of round pole lagging 6 to 8 inches in diameter was maintained over the manway side of the raise. This bulkhead was raised from time to time as the raise advanced. After the first 150 feet of raise was completed a heavy pole lagging bulkhead was placed over the manway side, and the raise continued by working on staging placed across temporary stulls. When all the ground was broken the dividing stulls were placed in position and the chute was lined. The method of timbering this raise is shown in Figure 7.

A small air tugger was used to hoist timber and supplies. A snatch block was fastened to the drilling platform. A fan operated by compressed air was used to ventilate the raise through a 10-inch pipe carried up the manway side.

The manway raises are carried up a short distance above the back of the stopes as the stopes advance. They consist of one compartment 4 by 4½ feet in the clear. The manways are lagged off from the broken ore by 2 by 12 inch plank nailed lengthwise to stulls placed on 5-foot centers. The ends of the stulls are elevated slightly above normal on the hanging wall side, so that the weight of the broken ore tends to tighten the stulls in place.

The chute raises are driven to a height of 25 feet above the floor of the level. Temporary staging of stulls and plank are used in drilling.

A raise crew consists of two men working on one shift. The two men do all the drilling and timbering; the ore broken is drawn from time to time by trammers. The routine of operations in advancing a raise is about as follows:

Day shift (2 miners)		Night shift (1 trammer)	
Operation	Time	Operation	Time
Barring down	30 minutes	Tramming when necessary	- -
Erecting staging	1 hour		
Preparing to drill	30 minutes		
Drilling	4 hours		
Removing equipment	15 minutes		
Loading	30 minutes		
Blasting	15 minutes		

Raise rounds are drilled with stoper machines of the self-rotating type, using 1-inch quarter octagon steel. The average raise round is shown in Figure 6. This round is varied, however, in the number and spacing of holes, according to the character of the ground. Gelatine dynamite of 40 per cent strength, detonated with No. 6 detonators and single-tape fuse, is used for blasting.

Digitized by Google

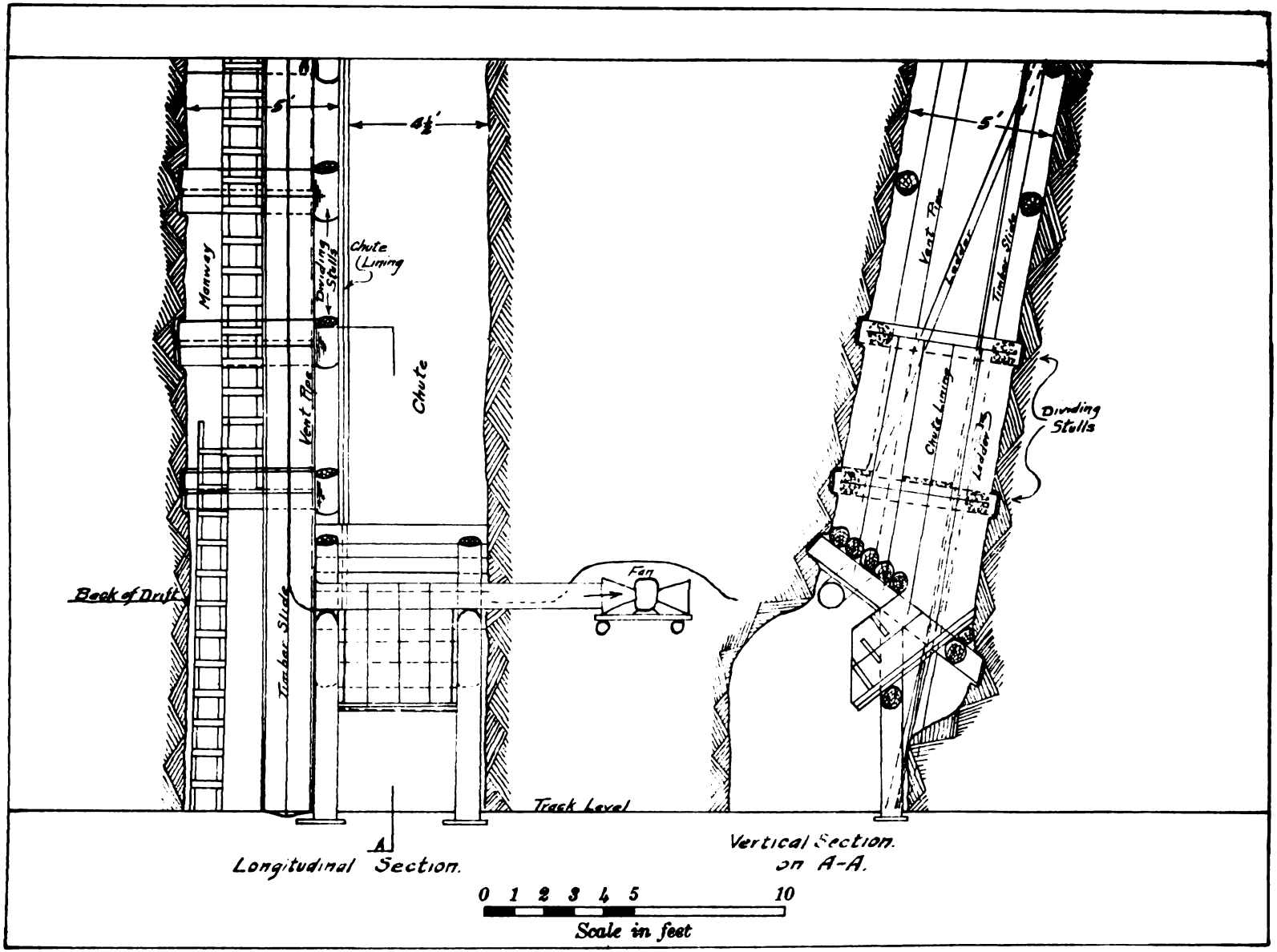


Figure 7. – Details of 2018 ore pass

Digitized by Google

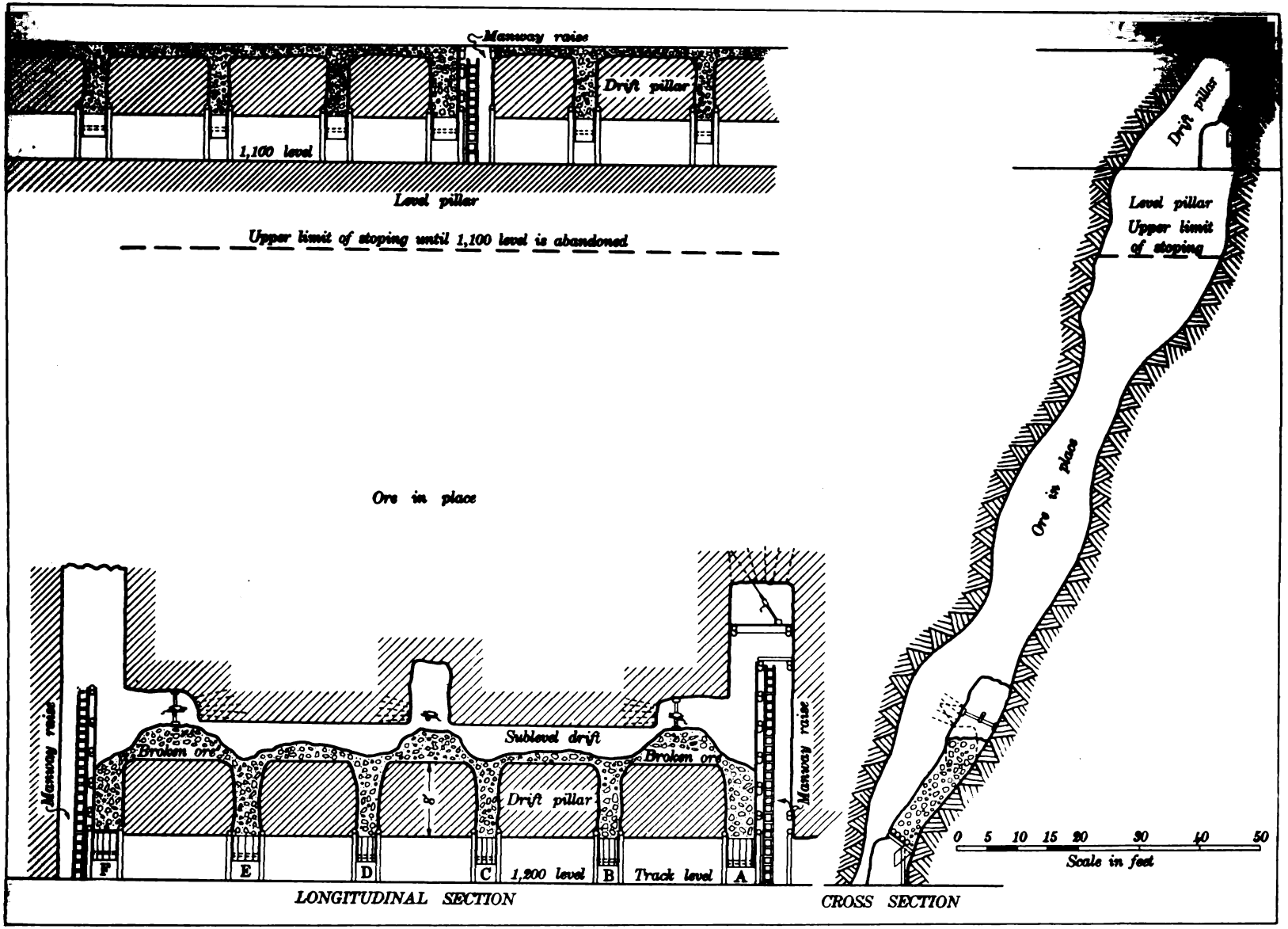


Figure 8. – Sections through 12-2115 stope, showing stope preparation complete and stoping operations just beginning

Digitized by Google

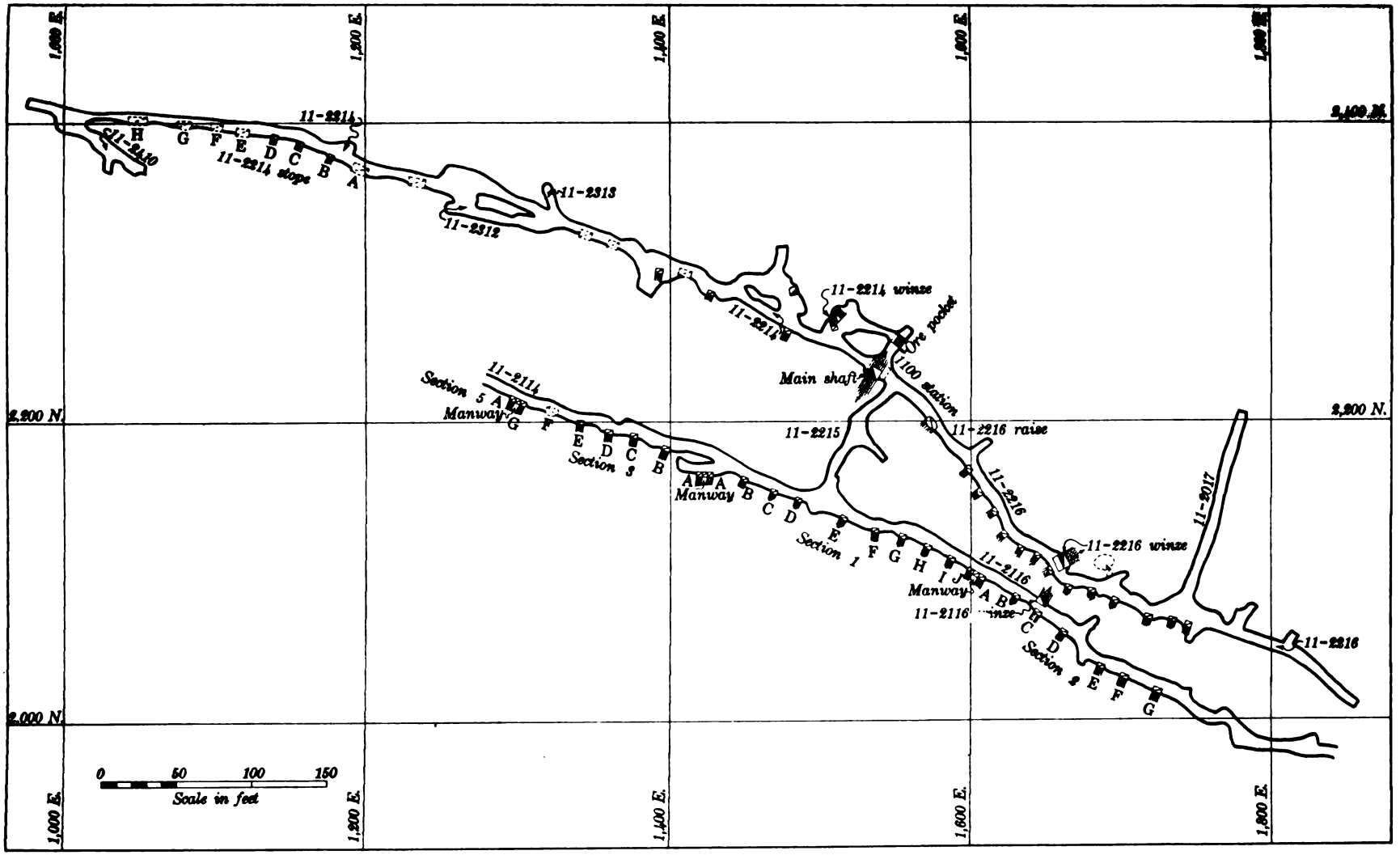


Figure 9. – Plan of 1,100 level, showing draw chutes from shrinkage stopes, and stope layout

Digitized by Google

STOPING

Figure 8 shows a longitudinal and a cross section through a completely developed shrinkage stope. The method used at Cortez is a shrinkage system which starts from a sublevel 15 feet above the track of the main level. By starting stoping operations from the sublevel the use of timber for supporting the ore above the drift is avoided and less shoveling is necessary.

Due to the irregular occurrence of the ore in the fissures, it has been found expedient to determine the extent of the ore in plan before starting stope development. The stope layout can then be planned to better advantage. In addition should this not be done a large amount of development work might be done to mine a small amount of ore. Figure 9 is the plan of the 1100 level.

Where the ore body is of sufficient length the stope is laid out in sections with manway raises spaced at intervals of 100 and 125 feet. The following description carries through the stope development of one such section of 12-2115 stope on the second level below the Arctic adit.

The main drift was driven along the strike of the ore body and the length of the shoot determined. At either end of the 120-foot section manway raises "A" and "F" (fig. 8), were advanced to a height of 25 feet above the track by working on temporary staging. The manway raises were then timbered with stulls placed on 6-foot centers. While this timbering was in progress the chute raises "B" and "E" were started, advanced 25 feet above the floor of the level, and the chutes installed. Chute raises "C" and "D" were next driven and the chutes built. At the same time that raises "C" and "D" were advancing a sublevel drift was started from the manways at both ends of the section. The sublevel-drift floor was carried at an elevation of 15 feet above the haulage level drift, leaving an 8 foot pillar to support the broken ore. The chutes are spaced at intervals of 24 feet.

Before the sublevel drift had been driven the entire length of the stope section, the manways "A" and "F" were advanced an additional 25 feet and timbered with stulls as before to a height of two sets above the back of the sublevel drift. This completed the preparatory work for stoping operations.

Stoping consisted of blasting horizontal rounds in the face of the stope. Slices were started at each manway and the miners advanced toward each other. When they met in the center of the stope another slice was taken in the same manner. This procedure was repeated until within 15 feet of the level above. The 15 feet of ore was left as a floor pillar. The manways were carried up from time to time as the stope was advancing. Sufficient ore was withdrawn from the stope to keep the miners within convenient distance of the stope face for drilling.

Occasional pillars of low-grade ore were left in a stope to support the hanging wall. Any pillars of good ore that were needed for support were blasted when the ore was being drawn from the completed stope.

Digitized by Google

The ore chutes are standardized. They are of simple design as shown in Figure 10. The gates are lengths of 2 by 12 inch plank which slide against wooden cleats nailed to the sides of the chute. Several inclinations of the chute bottom were tried, and it was found that with a steeper angle than 38° the ore pressed heavily against the chute gates so that it was difficult to raise them, and when once raised it was difficult to stop the flow of ore. With a flatter angle the ore would not run readily, especially when it was wet.

The advantages of the method of developing a shrinkage stope as practiced at Cortez is that stope preparation is expedited by the number of places that can be worked without interfering with each other; a maximum of the ore broken in developing a stope can be drawn directly into the cars without shoveling; the stope is fully developed before stoping operations are started so that the ore can be attacked in a systematic manner.

At Cortez the ore shoots are irregular in size, and sometimes changes are necessary in the mining method to suit the changing conditions.

From 30 to 35 per cent of the ore broken in stoping operations can be drawn off; the balance is left in the stope to provide a working floor. Close supervision is necessary to prevent the drawing of too much of the ore from the stope. When the ore is drawn down too far it is impossible for the miners to drill in the stope without employing temporary staging.

All supplies are carried by hand into the stopes. Natural ventilation of the stopes keep the working conditions excellent as a rule; only when one of the manways at either end of a stope section is blocked is the air condition bad.

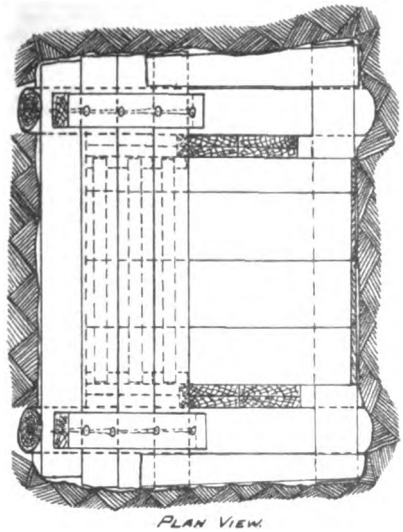
The type of round used is not standardized because of the varying conditions of width, character of the ore, and condition of the drilling face. Drilling is done with Leyner machines mounted on a crossbar where the width permits, or on an arm attached to a column which is set up on the broken ore.

It has been found that by using more holes and more explosive than are normally required to break a stope round that the ore is broken finer, obviating the necessity of plugging boulders in the stopes and preventing to a great extent the choking of the chutes.

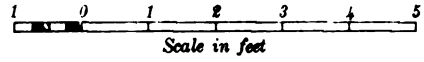
EXTRACTION

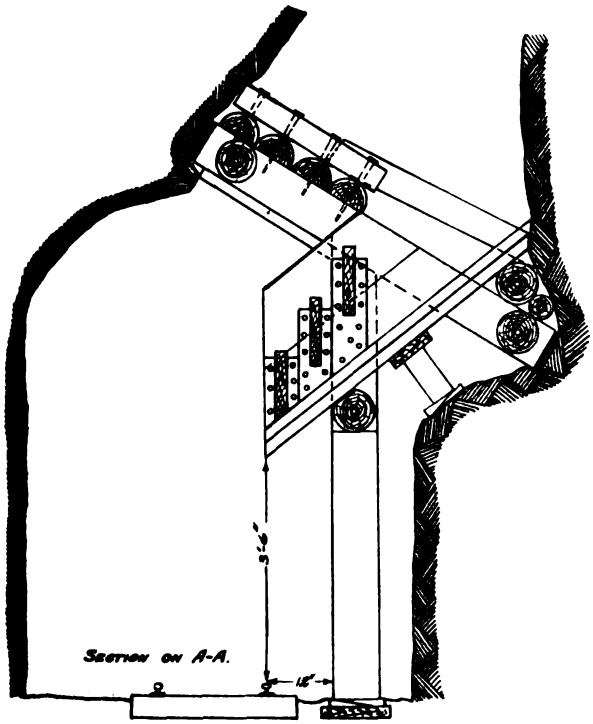
The percentage of ore extraction in mining by the shrinkage method as applied at Cortez is estimated at 95 per cent. The remaining 5 per cent represents the pillars which are left under the floors of the levels and above the drifts. These pillars, however, are not a total loss as they may be recovered eventually when the levels are to be abandoned. If the stopes have been filled with waste, as has been done in some cases, the drift pillars can not be recovered without excessive cost.

Digitized by Google



PLAN VIEW





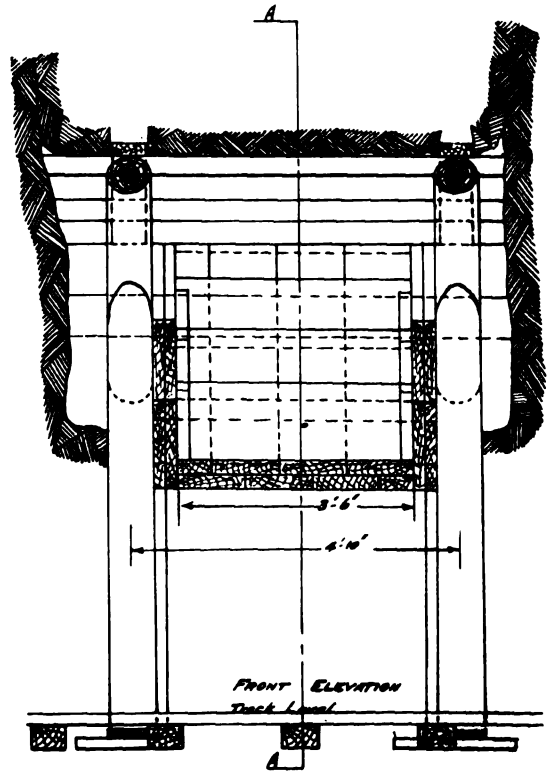


Figure 10. – Standard ore chute. Governing dimensions shown, al' o he· dimensions depend on how closely the ground is broken

Digitized by Google

It is hard to estimate the amount of dilution which takes place in mining, but it is known to be of considerable importance. In the shrinkage system of mining no sorting of waste from the ore can be done economically in the stopes. The causes of dilution in shrinkage stoping are as follows:

(1) Deliberate blasting of waste into the stopes, where it is necessary to prospect cross fissures not of mining width.

(2) Spalling of the wall rock, especially in proximity to cross fractures.

(3) Improper placing of holes and the blasting of wall rock in the ore.

This latter cause accounts for the major portion of the dilution. It can be reduced to a great extent by careful sampling, and by close supervision of the drilling and blasting.

APPLICATION OF THE SHRINKAGE SYSTEM

In mining by the shrinkage stope method at Cortez each stope presents its own problems, so that no definite standardization of methods can be adopted. The necessity of maintaining the daily tonnage for the mill has a bearing on stoping operations, as it is sometimes difficult to build up the reserve of broken ore in the stopes and still maintain the production. In a large operation this factor is not so pronounced as in a small mine like the Cortez.

The shrinkage method of mining is applicable when (1) the angle of repose of the broken ore is less than the angle of dip of the footwall, so that the ore will run into the chutes without shoveling; (2) the walls of the ore body are firm and stand well with little support; (3) the ore can be mined without the necessity of sorting; (4) the ore body is sufficiently large to produce the tonnage for the mill by taking only one-third of the ore broken.

The latter factor in mining by the shrinkage method can be also obtained if there are a number of small ore bodies which can be mined at one time.

The advantages in mining by the shrinkage method are as follows:

(1) Cheapness: no shoveling is necessary, very little timber required, no waste filling necessary.

(2) Very small amount of preparatory work required.

(3) It is easier to follow the ore when the walls are poorly defined by mining by overhand stoping than in any other direction, as the gradation between ore and waste can be more readily discerned.

Digitized by Google

(4) Safety; the men are always working close to the back so that close inspection can be made, reducing the danger of missed holes and falling rock.

The disadvantages are as follows:

(1) A large amount of broken ore, representing considerable potential capital is tied up in the stopes for long periods. The length of time this capital is tied up depends on the rapidity of mining and the distance between the levels. The greater the distance between levels the longer the time required in completing the stope, and the larger the amount of ore in the stope.

(2) Where the ore bodies are small a large number of places of attack must be mined simultaneously as only one-third of the broken ore can be drawn until the stope is completed to the top of the ore body or to the level above.

(3) Prospecting can not be done to advantage by driving crosscuts into the walls of the stope while stoping progresses without diluting the ore already broken in the stope.

(4) No grading or sorting can be done in the stopes.

The best plan with a shrinkage system of mining would be to have at least three stopes going at the same time; one stope completely mined, ready to draw, to provide most of the daily mill tonnage; the second in the stoping stage, which can furnish part of the required tonnage as represented by the increase in volume of broken ore over ore in place; the third in the development stage. Exploration work should be far enough in advance to insure this condition continually.

TRANSPORTATION

Tramming from the stopes to the shaft ore pockets on the two lower levels is done by hand with 18-cubic-foot capacity, flat-bottomed, end-dump, cars equipped with 10-inch diameter, 3-inch treadcast iron wheels with roller bearings. The track gage is 18 inches; rails weigh 20 pounds per yard, and ties are 4 by 6 by 30 inches long spaced about 30 inches apart. Switches are homemade, with no attempt at standardization.

Hoisting from the two lower levels is done in an 18-cubic-foot capacity self-dumping skip. The skip has 8-inch diameter roller-bearing wheels with a wide tread on the lower wheels for dumping the skip. The hoisting cable is $\frac{3}{4}$-inch, 6-strand 19-wire plow steel fastened to the skip with Crosby cable clamps. The head sheave is 5 feet in diameter, made of cast iron with wood fillers in the groove. The hoist is of the single drum type with a single gear reduction and is driven by a 50-horsepower, 2200-volt, 3-phase, 60-cycle induction motor.

Digitized by Google

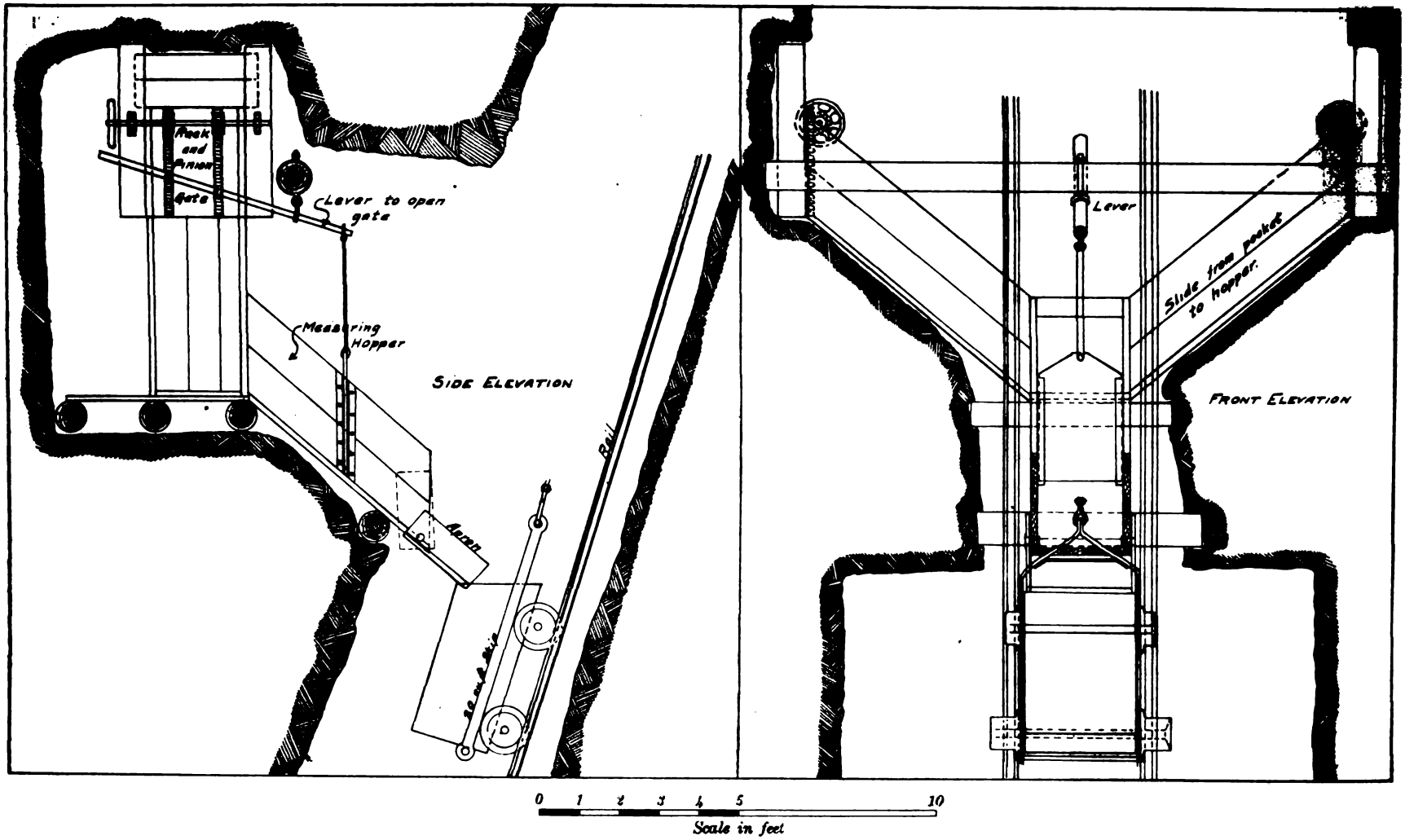


Figure 11. — Detail of 1900 loading pocket

Digitized by Google

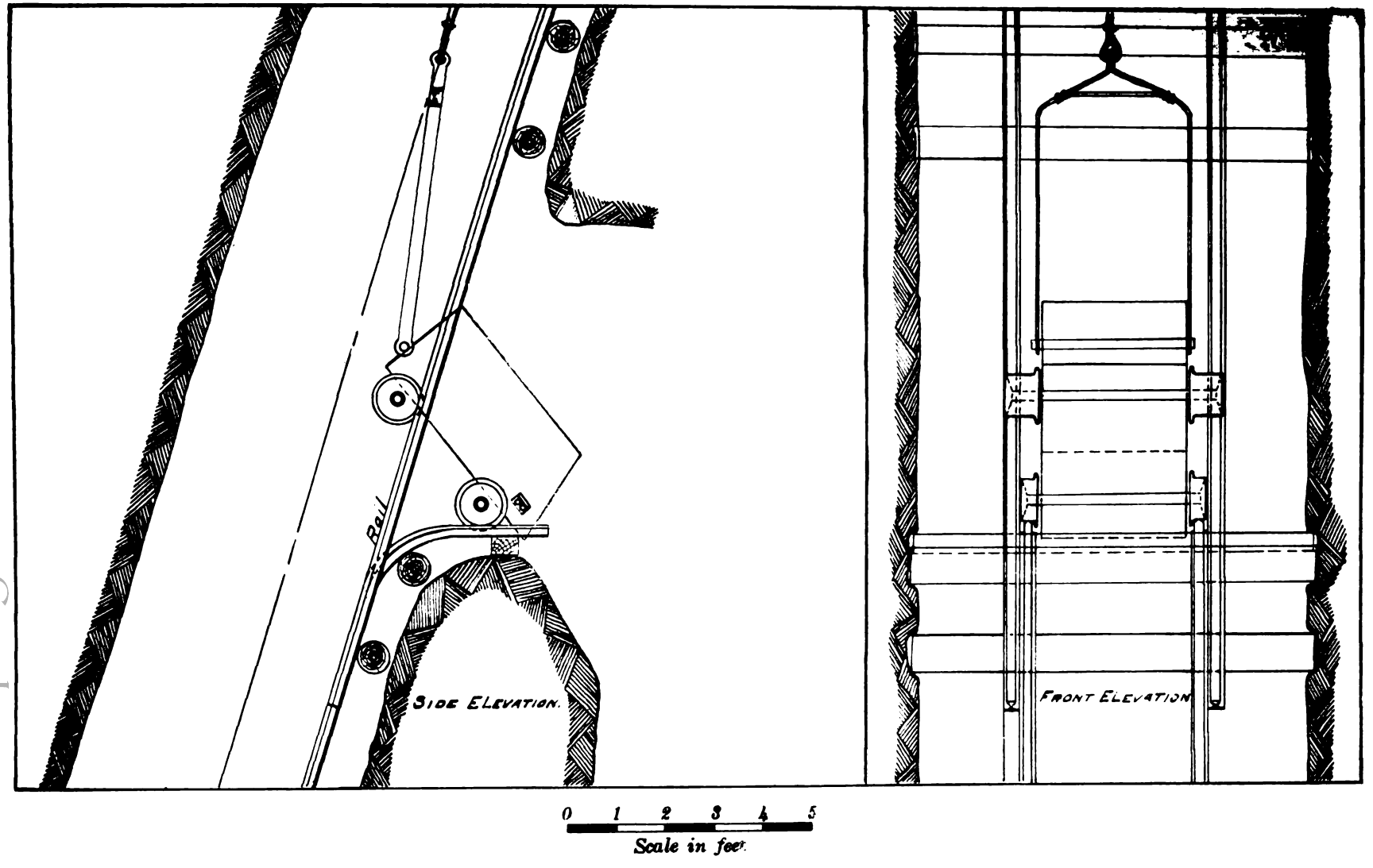


Figure 12. — Details of dumping device. Skip in dumping position

Digitized by Google

Figure 11 shows the loading pocket on the bottom level. Figure 12 shows details of the dumping device with the skip in dumping position. Figure 13 shows the shaft layout, showing skip loading arrangement on each level and dumping pocket on the haulage level.

The distance from the ore pocket on the adit level to the portal of the adit is 3,200 feet. Hauling on this level is done with a 3½-ton storage-battery locomotive. The locomotive is equipped with a spare set of batteries, so that one set of batteries is being charged while the other is in use. Each set of batteries has a capacity of 20 kilowatt hours, and a fully charged set will handle 45 to 50 cars in a shift without recharging. The change of batteries is made with a patented inclined transfer rack.

Cars on the adit level are of the gable-bottom, side-dump type having a capacity of 50 cubic feet or 6,000 pounds of ore. Wheels are of cast steel, are 14 inches in diameter, and have a 4-inch tread and roller bearings. A train consists of 6 cars. A motorman and a helper comprise the train crew and do all the loading at the chutes and dumping at the mill; they also handle all material and supplies taken into the mine.

The main haulage track has 30-pound rails, 4 by 6 inch ties, and a 24-inch gage. The track grade is about ½ of one per cent.

WAGE SCALE

Practically all work is done on company time. Long raises or development drifts are sometimes driven on contract. The contract price is fixed for each job and varies with the working conditions.

Underground employees are comprised of about 75 per cent native Americans, 15 per cent Mexicans, and the remainder of various European nationalities. Due to the fact that the district is isolated and accessible only by automobile, labor is difficult to secure. The wage scale as of December, 1929, for the principal classifications of employees was as follows:

Blacksmith	$7.00
Blacksmith's helper	5.75
Hoist engineer	6.00
Mucker	5.25
Machineman (miner)	5.75
Motorman	5.75
Motorman's helper	5.25
Pipe and trackman	5.75
Shaftman	6.25
Timberman	5.75
Trammer	5.25

Digitized by Google

VENTILATION AND FIRE HAZARD

Ventilation is natural except in dead ends, where fresh air is furnished to the working faces by means of small blowers operated by compressed air, discharging through 10-inch galvanized iron pipe hung in sections 12 feet in length from sprags across the back of the drift or crosscut.

The fire hazard in the mine is almost negligible as very little timber is used in the mine beyond that required for chutes and manway timbering in the stopes.

GENERAL

Power is generated at the mine by two 200-horsepower semi-Diesel engines, burning 27 plus fuel oil, and each operating a 3-phase, 60 cycle, 460-volt generator. Fuel oil costs 9 cents per gallon at the mine, and power cost is about 1.10 cents per kilowatt hour.

Compressed air is furnished by 3 two-stage compressors. The main air line into the mine is 4-inch pipe leading into a receiver placed about 3,000 feet within the portal of the Arctic adit. Leading out of the receiver is a 3-inch pipe to the take-offs at the stopes.

Water for the mine and mill is pumped against a 500-foot head from wells in the valley about 3 miles south of Cortez. Domestic water comes from a spring 7 miles south of Cortez and flows by gravity to the camp.

Table 1. - Summary of costs

Cortez Mine — Year, 1929
Mining method: Shrinkage — Tons mined: 43,806

Underground costs per ton of ore delivered to the mill

	Labor	Supervision[1]	Compressed air, drills and steel[2]	Power	Explosives	Timber	Other supplies	Total
Development[3]	$0.774	$0.061	$0.180	$0.046	$0.191	$0.036	$0.139	$1.427
Mining	1.082	0.086	0.250	0.070	0.304	0.054	0.126	1.972
Transportation[4].....	0.410	0.032	- -	0.111[5]	- -	- -	0.025	0.467
Totals	$2.266	$0.179	$0.430	$0.116	$0.495	$0.090	$0.290	$3.866

Note: General underground expense and surface expense directly applicable to underground operations are distributed in the above accounts.

1 Supervision includes ½ superintendent and shift bosses.
2 Includes cost of compressed air, drills, and steel, steel-sharpening shop cost, drill repairs, and maintenance and installation of pipe.
3 Development not segregated in ore and rock.
4 Transportation includes hoisting, motor haulage, and hand tramming.
5 Power cost for transportation apportioned in development and mining costs. The figure shown is total power cost less cost of lighting, and it is not included in the totals.

Digitized by Google

Table 2. - Summary of costs in units of labor, power, and supplies

Cortez Mine			Year, 1929
Tons of ore delivered to mill: 43,806		Mining method:	Shrinkage
	Development	Mining	Total
A. Labor (man-hours per ton)			
Breaking (drilling and blasting)	0.508	0.901	1.409
Timbering	0.073	0.209	0.282
Shoveling	0.555	0.475	1.030
Tramming (hand)	- -	- -	0.269
Hoisting	- -	- -	0.086
Motor haulage	- -	- -	0.176
Supervision (shift bosses only)	- -	- -	0.110
Total labor underground			3.362
Average tons per man per shift			2.405
Labor, percentage of total cost			71.98
	Development	Mining	Total
B. Power and supplies			
Explosives (pounds per ton) 40 per cent gelatin, size 1-1/8 by 8 inches	1.27	2.03	3.30
Timber (bd. ft. per ton)	0.91	1.36	2.27
Total power (kw. h. per ton)	- -	- -	31.767
Air compression	8.875	12.304	21.179
Hoisting	3.740	5.631	9.371
Motor haulage	0.371	0.559	0.930
Lighting	- -	- -	0.285
Other supplies, in percentage of total power and supplies			35.16
Supplies and power, percentage of total cost			28.02

Digitized by Google

Table 3. - Details of costs in units of labor, power, and supplies

	Shafts	Drifts	Crosscuts	Raises
Size of excavation (feet)...	6 by 12	6 by 7½	6 by 7½	5 by 10
Timbered or not	Partly	None	None	Partly
Character of rock	Hard	Hard	Hard	Hard

Total feet of development work (not segregated) 3,077

A. Labor (man-hours per foot)

Breaking (drilling and blasting).....	7.243
Timbering	1.041
Shoveling and tramming	7.906
Hoisting	0.932
Motor haulage	1.051
Supervision	0.875
Total labor	19.048

B. Power and supplies

Explosives (pounds per foot) 40 per cent gelatin, size 1-1/8 by 8 inches	18.18
Timber (bd. ft. per foot)	19.31
Total power (kw. h. per foot)	134.8
Air compression	126.3
Hoisting	53.2
Haulage	5.3
Total	134.8
Other supplies, percentage of total cost	8.64

C. Percentage of total cost

Labor	71.33
Power and supplies	28.67

Digitized by Google

Table 4. - Details of development costs per foot

	Shafts	Drifts	Crosscuts	Raises
Size of excavation (feet)...	6 by 12	6 by 7½	6 by 7½	5 by 10
Timbered or not	Partly	None	None	Partly
Character of rock	Hard	Hard	Hard	Hard

Total feet of development work (not segregated) 3,077

A. Labor (cost per foot)		
Breaking (drilling and blasting)		$5.206
Timbering		0.716
Shoveling and tramming		5.188
Hoisting		1.637
Motor haulage		1.725
Supervision		0.875
Surface labor		1.029
Total labor		$16.376
B. Power and supplies (cost per foot)		
Explosives		2.727
Timber		0.520
Power		2.049
Air compression	1.389	
Hoisting	0.585	
Haulage	0.058	
Lighting	0.017	
Total cost of power	2.049	
Other supplies		1.999
C. Cost per foot		
Labor		$16.376
Power and supplies		7.295
Total cost per foot		$23.671

Digitized by Google

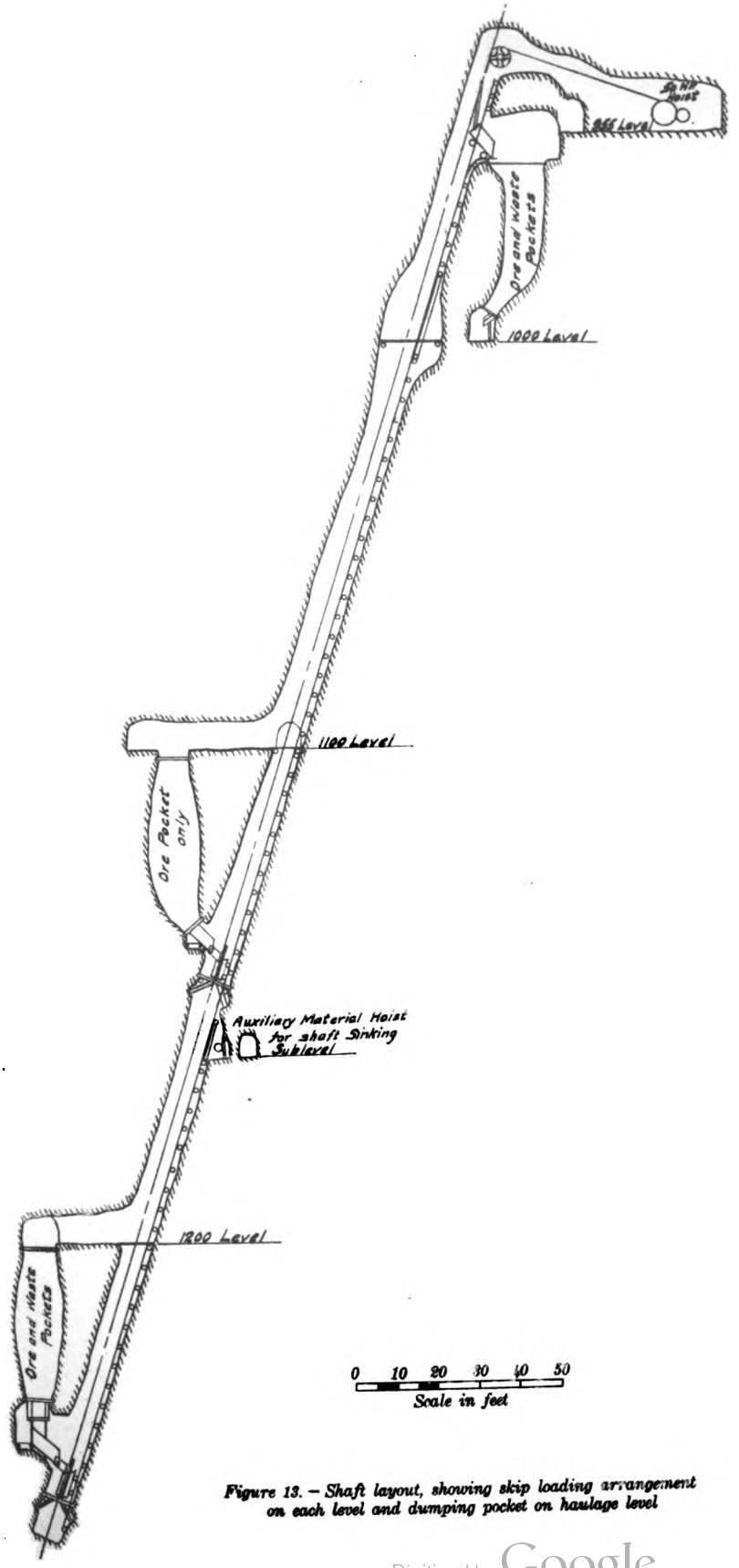


Figure 13. – Shaft layout, showing skip loading arrangement on each level and dumping pocket on haulage level

Digitized by Google